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MINING METHODS AND COSTS AT THE PORCUPINE UNITED GOLD MINES' ROCHESTER MINE AT TIMMINS, ONTARIO[1]

By J. D. Tolman[2]

INTRODUCTION

In view of present interest in a revival of gold mining enterprises, papers dealing with the development of small gold properties are apropos. The present paper deals with the operation of a small gold property in the exploration and development stage, mining and milling 25 tons of ore daily, and indicates what may be expected in the way of mining costs under northern climatic conditions, by practicing rigid economy, at a property where disproportionate overhead costs are unavoidable bacause of the small rate of production. The output of this prospect is treated in a small amalgamation plant which has been described in a previous circular.[3]

HISTORY

Prospecting in the district was begun in 1909, and by 1911 the adjoining mines, Hollinger and McIntyre, were in operation. Actual work was begun on the Rochester claim in July, 1922, and carried on through 1924. The property was closed until 1928 when the present company took it over, together with the other holdings of the Canadel Mining Co. A small mill was started as a pilot plant on March 24, 1929, and is still in operation.

GEOLOGY

The rocks of the district are pre-Cambrian, and consist of basalt and greenstone of the Keewatin series, with an area of basaltic schist and greenstone schist lying between the two. Intruding into the basaltic and greenstone schist is a quartz feldspar porphyry. The porphyry has probably had some influence on the gold-bearing veins, as they are found in the schist in the vicinity of the porphyry.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

The ore consists of quartz with mineralized stringers of basalt running through it, and of banded quartz and schist, or of stringers of quartz in the schist with considerable pyrite, the best values being where there are stringers of very fine pyrite, sometimes accompanied by visible gold.

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1 The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6470."

2 Superintendent, Porcupine United Gold Mines.

3 Vary, Ronald A., Amalgamation Practice at Porcupine United Gold Mines, Ltd., Timmins Ontario: Inf. Cir. 6433, 1931, 5 pp.

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The ore body now being worked, which is one of several lenses disclosed on the property, varies in width from about 2 feet of solid quartz with mineralized basaltic stringers containing coarse and fine pyrite. The ore occurs in lenticular bodies, sometimes narrowing to a few inches in width and then widening to 6 feet or more. In following the vein from bottom to top the ore steps off at intervals into the footwall. The values are somewhat irregular and samples taken across the vein vary considerably, but values in this vein are fairly high for this district.

Both the footwall and the hanging wall have been subject to shearing and scale off in large slabs. This caused great dilution in shrinkage stoping without timbering, but timbering in the present stopes will remedy this to a large extent.

METHODS OF PROSPECTING AND EXPLORATION

The property was formerly quite extensively diamond-drilled. A two-compartment shaft was sunk to a depth of 947 feet, and stations were cut at the 100, 300, 500, 800, and 900 foot levels. Most of the veins indicated by the diamond drill have been cut and drifted on.

The vein now being worked was drifted on, and some short crosscuts were driven. Test holes are now put in the stope walls at about 10-foot intervals and each 2 feet of cuttings is assayed.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGE AND VALES

Face and back samples are taken in drifts; breast and back samples are taken in stopes at about 5-foot intervals, and test-hole samples are taken in the walls of the stope. Assay plans are made of drifts, and longitudinal sections are made of stopes, showing each sample number in place and a table at the side gives widths and values for the computing of values in the stopes. High assays are reduced in calculations.

Ore in place is computed at 12 cubic feet per ton and broken ore at 20 cubic feet per ton.

METHODS OF DEVELOPMENT AND MINING

The shaft is vertical to a depth of 947 feet and there are levels at 100, 300, 500, and 900 feet. A station is cut at 800 feet, but very little work has been done on that level.

So far the only stoping has been done on the 300 and 500 foot levels. The 500 was stoped through to the 300 in one continuous stope.

The shaft is a two-compartment shaft, a cageway and a manway. Each compartment is 4 feet 6 inches by 4 feet inside, and 10 by 10 inch timbers are used.

Crosscuts are untimbered. The timbers in the drifts are 12 to 14 inch stulls, good deep hitches being used, and posts are put in at the chutes. The bottoms of the chutes are of 6-inch lagging, inclined at 35°, and two sets of rails are used for supporting the plank and the stop logs at the chute mouth.

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Drifting was done with Ingersoll-Rand R-72 drill and with Canadian Pneumatic No. 6 drill using 1-inch, quarter-octagon steel. A diamond or sometimes a V cut is used. The cut is blasted first and the remaining holes later. The cross section is about 5½ by 7½. Seventeen holes are required for breaking a round on the upper levels, and the harder formation below requires 21 holes. On the upper levels a round per shift was made, but below it was cheaper to put in a slightly longer round and blast every other shift.

Raises are driven with R-51 stoper machines, and any long raise is usually let by contract.

Stoping

Shrinkage stoping was used on the 500-foot level and then the stope had to be scaled as drawn and followed down by timbering with stulls, because both foot and hanging were inclined to slab off. A great deal of dilution occurs in this way and in the stope worked at present, care is taken to break narrow, to the exact limits of the vein as it appears. Test holes are put in the walls and stull timbering is carried up with the stope, In this way dilution will be largely avoided. Pillars were left in the upper portion of the lower stope and are also being left at intervals in the present stope. Leaner portions are selected for the pillars.

In starting the stope, the back is taken down to 16 feet in height, stulls put in with posts where necessary, chutes put in at about 25-foot intervals and lagging put in on top of the stulls. Enough ore is drawn from the stope to keep the back at a convenient height for drilling (about 8 feet), and the ore is left in until the stope reaches the level above, when drilling in the stope is finished and all the ore can be drawn.

Transportation

The cars used are roller-bearing dump cars of 18 cubic foot capacity. The trammers from the chutes load in pairs and so can handle the planks at the chute mouth, which rest on rails, held by staples on the posts and caps at the side of the chute. The ore is all sorted at the chute by the trammers, since it is necessary to maintain a good grade of ore due to the fact that the amalgamation process is used in the mill. The slabby character of the wall rock, which is schisted, makes it easy to identify. On the 300 level the waste is dumped to fill up an old stope from the 500 level, but below the 300 level the waste had to be hoisted to the surface; it was often necessary to pile this waste at the side and hoist when convenient, as the hoisting capacity at times precluded the hoisting of waste except at irregular intervals. All fines are sent as ore, and although this causes some dilution by reason of the breaking up of the schist, the mineralized schist breaks up more readily and the greater proportion of the fines should be ore. Sometimes the waste sorted out runs as high as 40 per cent of the total rock broken, but the narrow timbered stope now being worked gives about 10 to 15 per cent waste. Two men load the cars and tram to a switch, and another trammer picks up the loaded car and leaves an empty, tramming the loaded car to the shaft, where it is loaded on the cage, hoisted to the surface and dumped on a grizzly. Any noticeable waste is sorted out at this grizzly by the top man, and the man at the crusher also throws out a small amount, although more than 90 per cent of the sorting is done underground.

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COMPARISON OF VARIOUS METHODS USED IN DISTRICT

Shrinkage stoping has been used by all the mines in the district to a great extent. As the larger mines have reached a greater depth pressure has apparently increased and the walls have become more difficult to hold. This has resulted in the use of square-setting in some stopes, and the cut-and-fill system is now being used in some of the larger mines. The principal reason for using the cut-and-fill system, however, is that it gives better opportunity for exploring the walls of the stope with safety. It is more expensive than shrinkage stoping, but it is thought that the ore gained more than compensates for the added expense. Shrinkage stoping is still used in the smaller mines and on the upper levels. No other method has been tried in the Rochester.

WAGE, CONTRACT, AND BONUS SYSTEM EMPLOYED

Crosscutting and drifting were done on a contract system, but all work at present is done at day's wages and no bonus system is used. All except the shift boss are paid by the hour.

The rates are as follows:

Shift boss	$7.00 per day
Timbermen and drill runners	.60 per hour
Muckers and helpers	.53 do.
Pumpmen, who are also deckmen	.60 do.

VENTILATION

The ventilation system is natural, and when the air is turned on after blasting the smoke clears in a short time.

FIRE HAZARDS

The main shaft is dripping wet and the timber in the mine is damp. Old timber is hoisted to the surface and all powder boxes and chips are taken out or hoisted with the waste. Hose and fire extinguishers are available.

SAFETY METHODS AND FIRST-AID ORGANIZATION

The mine is located practically in the town of Timmins, where the Government maintains a mine rescue station. A first-aid kit is kept at hand in a room adjoining the office of the company and is available at all times. No organized crew is trained for mine rescue work.

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Table 1.- Underground costs per ton of ore hoisted[1]

Rochester mine, April, 1929, to August, 1930, inclusive.
Tons hoisted, 10,000.

	Labor	Supervision	Air, drills and steel[2]	Power cost	Explosives	Timber	Other supplies	Total
Development	$0.096	$0.026	$0.055	--	$0.025	--	--	$0.202
Stoping	1.437	.347	.714	--	.305	$0.120	--	2.923
Tramming and hoisting	1.652	--	-- (hoisting)	$0.649	--	--	--	2.301
General underground expense	.385	--	--	.250	--	--	$0.207	.842
Surface expense applicable to underground	.028	--	--	--	--	--	--	.028
Total	3.598	.373	.769	.899	.330	.120	.207	6.296

1 On account of sorting out waste at the chutes, about 35 per cent more rock was broken than was hoisted. Some of this waste was hoisted and some used as filling in stopes. Above figures are based upon tons of ore actually hoisted.

2 Includes power for air compression.

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Table 2.- Summary of costs, in units of labor, power, and supplies

Porcupine United Gold Mines, April, 1929 to August, 1930.
Tons, 10,000.

	Man-hours per ton		
	Development	Mining	Total
A. Labor:			
Breaking (drilling, blasting, scaling)	0.112	1.536	1.648
Timbering	0.000	0.944	0.944
Mucking, haulage, hoisting	0.062	1.304	1.366
Supervision	-	0.368	0.368
General	-	0.384	0.384
Total labor underground	0.174	4.536	4.710
Average tons per man per shift			1.7
Labor, per cent of total cost			71.0
Average tons per man-shift on surface properly chargeable to underground			5.0
B. Power and supplies:			
Explosives, pounds per ton		0.926	
Timber, board feet per ton		2.666	
Total power per ton hoisted, kilowatt hours		91.6	
Air-compressor	26.0		
Hoisting	57.0		
Pumping	8.1		
Heating and Ventilation	0.3		
Lighting	0.2		
Other supplies, in percentage of total supplies and power		12.81	
Supplies and power, percentage of total cost		24.9	
Supervision, percentage of total cost		4.1	

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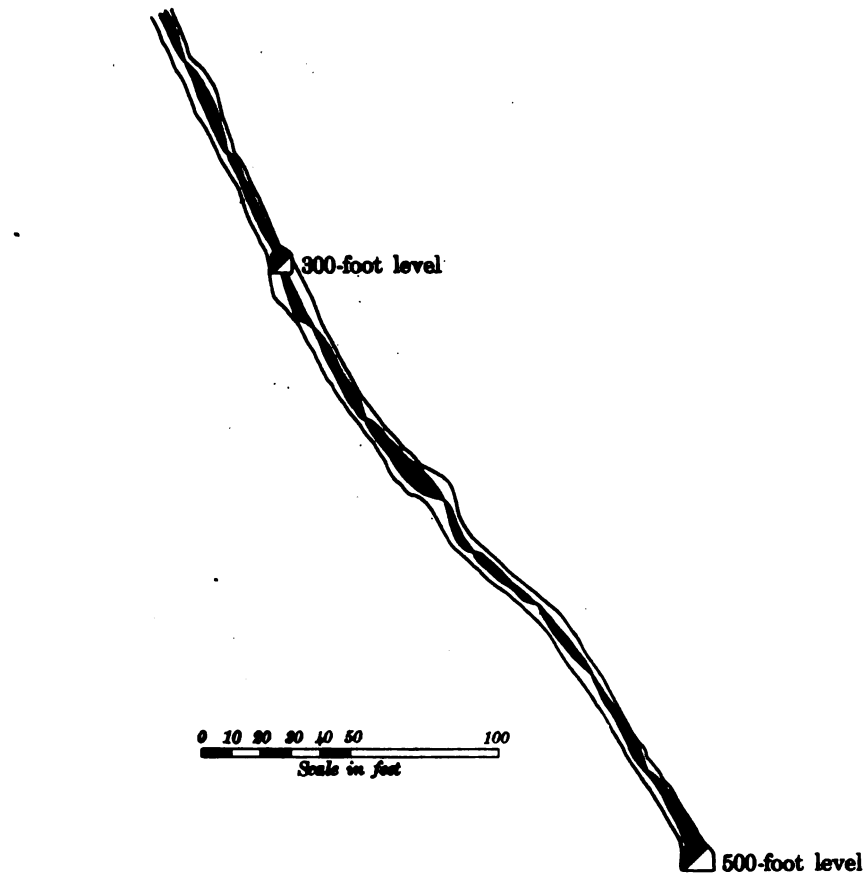

Figure 1.—Cross section through No. 10 vein. Looking east

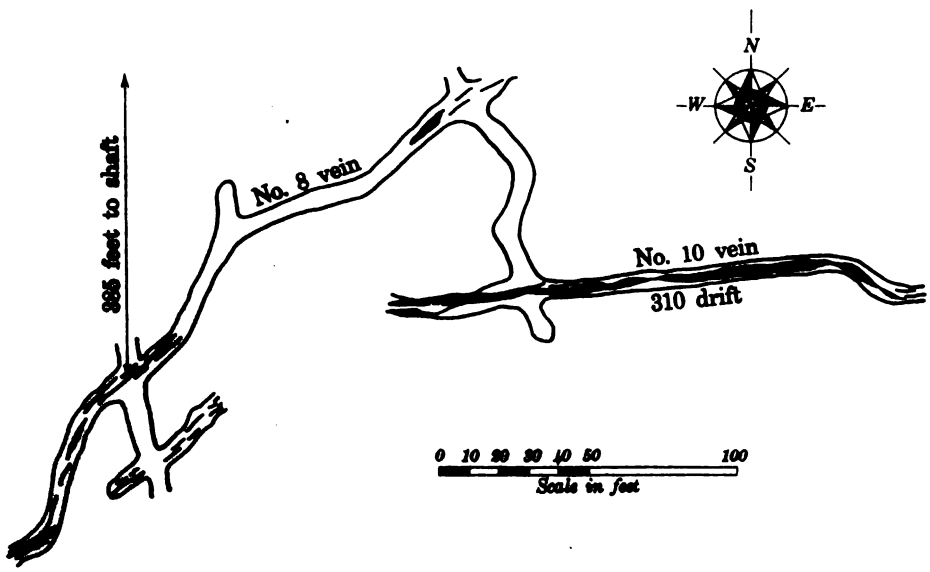

Figure 2.—Plan of No. 10 vein, 300-foot level

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